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DECEMBER, 1930

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METHODS OF MINING AT THE BLACK ROCK MINE, BUTTE & SUPERIOR MINING CO., BUTTE DISTRICT, MONT.

BY

D. B. MCGILVRA AND A. J. HEALY

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METHODS OF MINING AT THE BLACK ROCK MINE, BUTTE & SUPERIOR MINING CO., BUTTE DISTRICT, MONT.[1]

By D. B. McGilvra[2] and A. J. Healy[3]

INTRODUCTION

This paper, describing the mining practices and methods at the Black Rock mine, is one of a series of similar papers on mining methods in the various mines of the United States being prepared by the Bureau of Mines.

The Black Rock mine is located in the northeast corner of the Butte district, about 3 miles from the center of the town of Butte, and is owned and operated by the Butte & Superior Mining Co.

The district is accessible by four railroads, daily aeroplane passenger service between the towns of Salt Lake City and Great Falls, and improved automobile roads from various points.

Formerly, with large ore reserves, the Black Rock mine produced about 1,200 tons of ore daily and employed about 700 men. Production has decreased to about 500 tons per day at the present time. The ore is mined by a square-set-and-fill method and is shipped to a concentrator at Anaconda for milling. The principal mineral produced at present is zinc. Up to January 1, 1930, the mine has produced 6,363,316 tons of ore containing 1,725,704,828 pounds of zinc, 150,416,291 pounds of lead, 65,640,876 pounds of copper, 40,022,121 ounces of silver, and 58,365 ounces of gold.

GEOLOGY

The geology of the Butte district and of the mines has been described by the U. S. Geological Survey.[4]

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1 The Bureau of Mines will welcome reprinting of this article, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6370."

2 One of the consulting engineers, U. S. Bureau of Mines, and superintendent, Butte & Superior Mining Co., Butte, Mont.

3 One of the consulting engineers, U. S. Bureau of Mines, and chief engineer, Butte & Superior Mining Co., Butte, Mont.

4 Weed, Walter H., Geology and Ore Deposits of the Butte District, Montana, U. S. Geol. Survey, Prof. Paper 74, 1912, 262 pp.

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The Black Rock mine is on the eastern extremity of the Rainbow vein, which has a general east and west strike and an average dip of 80° to the south. The width of the vein ranges from a few feet to 120 feet. Stoping widths range from one to 12 sets, or 70 feet. The bulk of the production has been derived from this vein. Other veins occur, some of them of commercial importance. These veins are invariably narrow, averaging about 6 feet in width.

The country rock is granite, which has been intensely altered in proximity to the vein. The principal ore minerals are sphalerite and galena. The gangue consists mainly of altered granite, quartz, and pyrite. The vein structure presents a banded arrangement of ore and waste parallel to the walls.

PHYSICAL CHARACTERISTICS OF ORE AND INCLOSING ROCKS

The ore ranges from soft crumbly sphalerite to hard flinty material composed of quartz and sphalerite.

Both walls of the vein are usually talc seams, which cause considerable difficulty in mining operations. When a portion of the vein is undercut the block of ore above tends to slide downward on the slippery talc walls. Within the vein there are slips at right angles to the walls. When being stoped the ore often breaks off to these slips.

The granite country rock adjacent to the vein is altered, in many places to such an extent that it may be crumbled in the hand. The degree of alteration of the granite is more pronounced near the wider portions of the vein and becomes less intense as the distance from the vein increases.

In general, the ore and inclosing rocks are not strong enough to support themselves even over small spans. Artificial support is necessary as soon as possible after an opening is made.

METHODS OF PROSPECTING AND EXPLORATION

At present all prospecting is done by means of drifts and crosscuts. In the past, diamond drilling and deep-hole prospect drilling with heavy Leyner machines were tried.

The chief objection to long-hole drilling in the Black Rock mine is that the information gained is entirely negative. A 1-inch stringer of soft ore would salt the samples from the hole for the next 10 or even 20 feet of drilling. The only results that could be depended upon were in holes where no mineralization was encountered.

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The diamond drill has been used mainly in prospecting for new ore bodies. The chief objection to this form of prospecting is the rather high cost for the meager information gained. It is believed that most of the veins in this district are worthy of considerable lateral development which can not be done with the diamond drill as cheaply as by crosscutting and drifting.

The major problem in exploratory work is faulting. The search for faulted segments of ore bodies is simplified to a great extent by studying the detailed geological maps of the underground workings.

SAMPLING AND ESTIMATION OF ORE RESERVES

All development headings are sampled after each round is blasted. Stope breasts are sampled at each alternate square set. Samples are taken with a hand pick. Structural irregularities of each breast sampled are considered in taking the samples. Each band of ore is sampled separately, and the width of the bands of ore and waste is measured. The average grade of the ore, when broken, is calculated from the measurements and the values of the samples.

Ore reserves are estimated by multiplying the length, height, and average width for individual blocks as outlined by development work. The tonnage factor is 10 cubic feet for each ton of ore in place. In estimating positive and probable ore, the characteristics of each section of the vein, as determined by past mining operations, are taken into consideration.

DEVELOPMENT

Shafts

Entry to the mine is made by three vertical shafts--one main hoisting shaft and two auxiliary shafts used for ventilation and transportation of supplies. The level interval is 100 feet for the first 1,900 feet, and 150 feet to the bottom stoping level, which is 3,000 feet.

Each of the three shafts is divided into three compartments, two of which are 5 feet by 5 feet 2 inches in the clear and the other 3 by 5 feet in the clear. The smaller compartment is used for air and water lines, pump columns, and electric cables, and as a manway.

One of the auxiliary shafts is used exclusively for ventilation purposes and the other for lowering supplies into the mine. This supply shaft extends to the 2800-foot level. From the 2800-foot to the 3700-foot level the main hoisting shaft has been sunk with an additional "chippy" compartment for lowering supplies to the deeper portions of the mine. An auxiliary hoist is located on the 2800-foot level to serve the chippy compartment.

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In sinking the main shaft a crew of six men is employed on each of three shifts. A crew comprises four miners, one bucket dumper, and an engineer.

The cycle of operations in shaft sinking is as follows:

Operation	Time
Drilling, loading, blasting ...	8 hours
Timbering	4 to 6 hours
Shoveling	16 to 20 hours

Unmounted machines of the jackhammer type are used in sinking.

The round used in medium ground is shown in Figure 1. In harder ground the bottoms of the cut holes are brought closer together.

Blasting is done with 30 per cent strength gelatin dynamite and No. 8 electric detonators. A few sticks of 60 per cent dynamite are used in the bottom of each cut hole.

Round stulls are used for blasting logs and are chained to the wall plates.

Shoveling is done by hand, using two sinking buckets of 13 cubic feet capacity.

The wall and end plates of the shaft are of 12 by 12 inch timber. Dividers are of 10 by 12 inch timber. Sets are placed on either 4 or 5 foot centers, depending on the ground. In heavy ground cribbing is used with the halved method of framing at the ends of the wall and end plates.

Parts of the main shaft are in soft swelling ground, and jacket sets have been installed at these places. The manner of framing and position of the members of the jacket sets are shown in Figure 2. The advantage in the use of the jacket sets is that replacements or repairs can be made in the shaft timbering without interfering with hoisting operations through the shaft. In several instances 300-foot sections of the main shafts were retimbered at the rate of 4 feet per day without interfering with the daily production of 1,400 tons of ore and waste. Without the jacket sets a repair job of this kind would necessitate a shutdown.

Drifts and crosscuts

Three sizes of drifts and crosscuts are driven.

Drifts in ore are driven 8 by 10 feet in cross section and are timbered with sill sets (fig. 3B).

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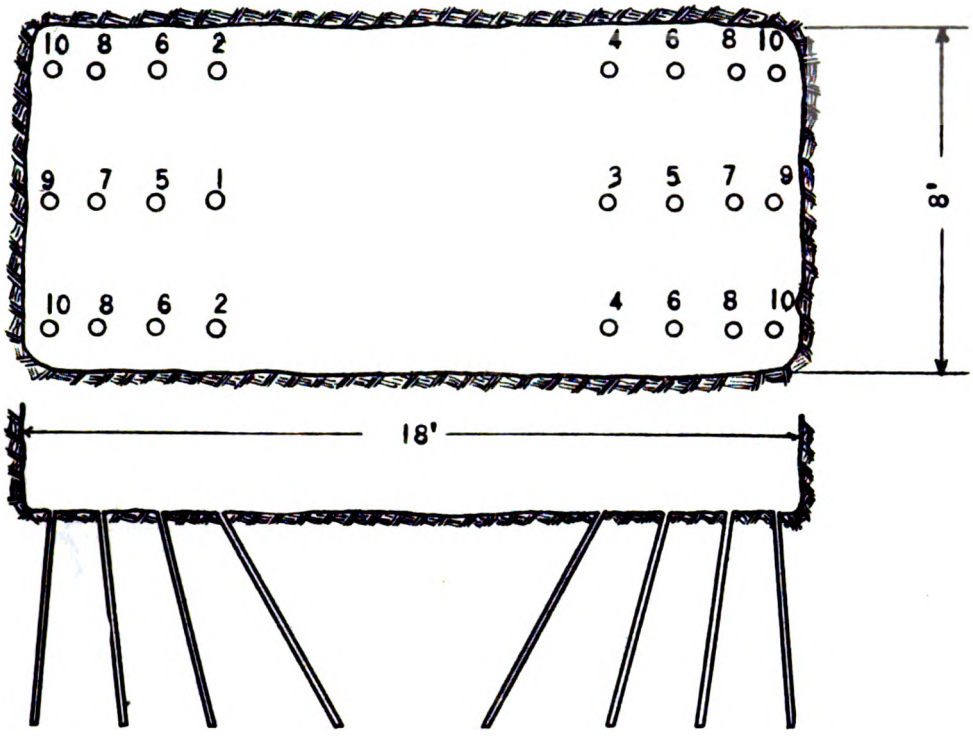

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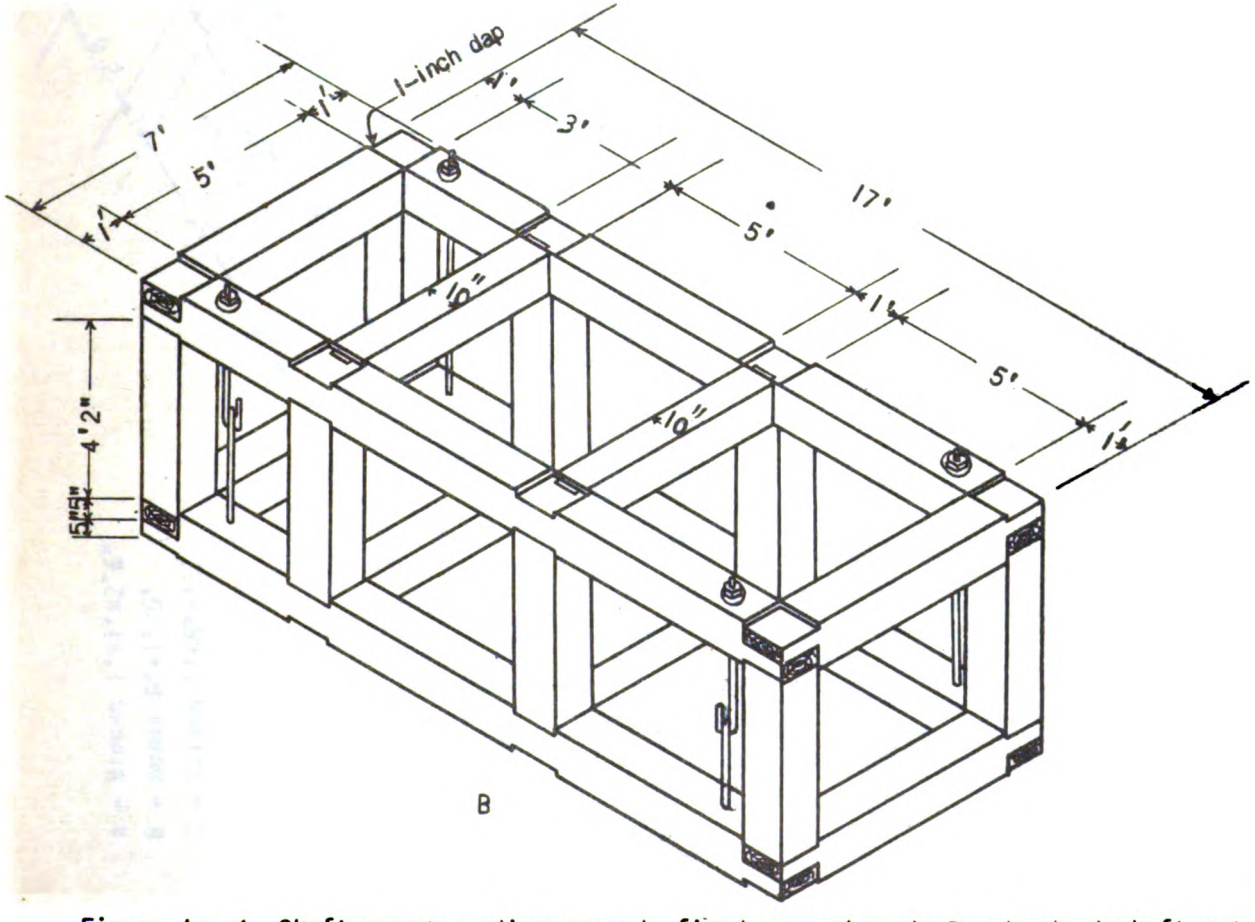

B

Figure 1.- A, Shaft round, medium ground, fired as numbered; B, standard shaft set

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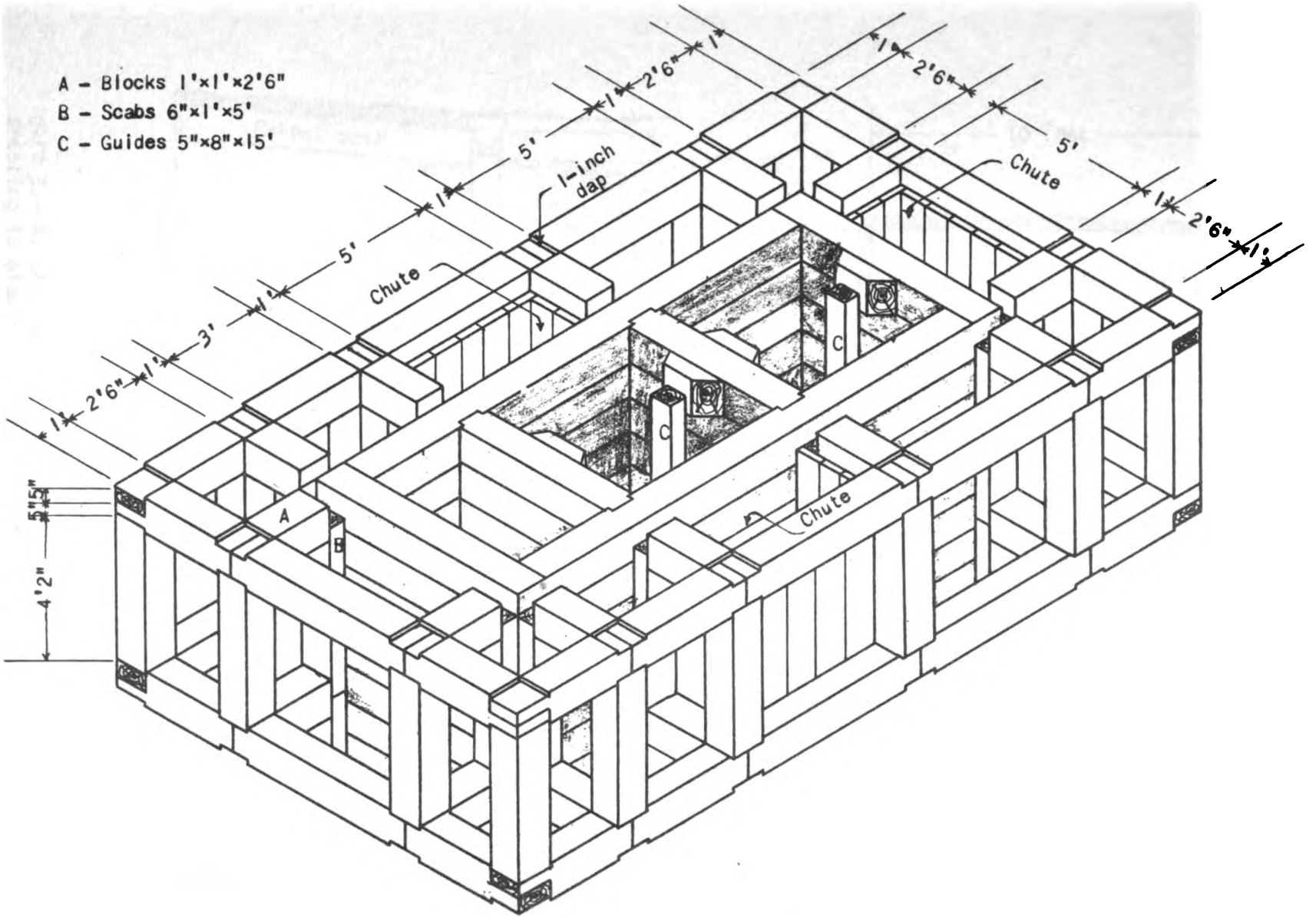

Figure 2.– Floor of jacket sets with cribbed shaft

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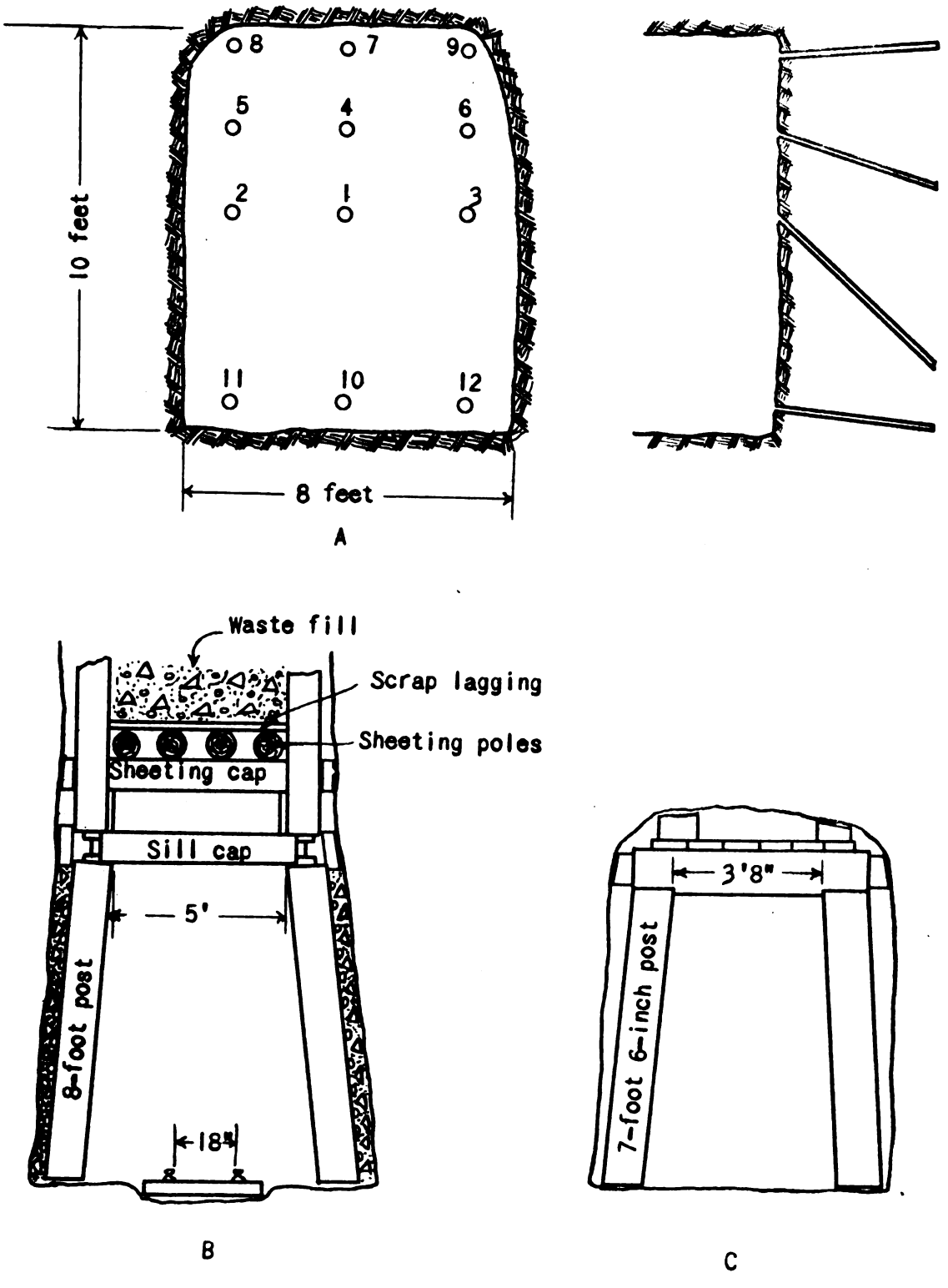

Figure 3.- A, Standard drift round in soft ground; B, sill set, sheeting in place; C, tunnel set

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Drifts or crosscuts driven in waste rock, where timbering is necessary, are driven 7 by 9 feet in cross section and are timbered with tunnel sets (fig. 3C). Very heavy drifts in ore are sometimes driven 7 by 9 feet and timbered with tunnel sets.

Exploratory crosscuts which do not require timber are driven with an arched back and are 6 by 8 feet in cross section.

Machines of the Leyner type, mounted either on a cross bar or on an arm attached to a column, are employed in drilling. The Hawkesworth bit[5] is used.

Ground conditions vary to such an extent that the employment of a standard round in drilling is impossible. The round used for drilling in soft ground is shown in Figure 3A. In medium hard ground a line of "easer" holes are drilled between the back and the breast holes. In drilling ravelly ore the round is drilled upward instead of down. When drifting is done with a smooth slippery wall on one side the cut is taken toward this wall. The average round will break about 4 feet ahead of the face.

Gelatin dynamite of 30 per cent strength is used throughout the mine. In places where the ground is tight three or four sticks of 60 per cent dynamite are used in the bottom of each cut hole and each lifter. The shift boss in charge of the men determines the amount of explosive to be used for each round. Blasting is done with No. 6 detonators and fuse. The primer cartridge is the second stick from the bottom in each hole. All rounds except shaft rounds are fired either with carbide lamps or a piece of lighted fuse. Stemming is not used.

Timbers are placed in position close to the working face, usually within two sets. In ground that sloughs readily the timber is placed up to the working face. In places it is necessary to use booms to suspend the cap and overhead lagging before mucking. After mucking is completed the posts are placed under the caps.

All shoveling is done by hand directly into cars. In driving a drift or a crosscut the usual arrangement is to employ two men on each shift. One crew of two men will drill, timber, and blast, and the following shift will shovel and lay track. When speed is essential three men are employed on each shift, and drilling and mucking are carried on simultaneously. The drill is mounted on a crossbar when driving a heading in this manner.

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5 Berrien, C. L., Development and Installation of the Hawkesworth Detachable Bit: Min. Cong. Jour., vol. 16, March, 1930, pp. 258-270.

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Raises

Raises are driven with a chute and manway compartment. The manway compartment carries the air and water pipes, ladders, timber slide and sometimes ventilation tubing (fig. 4B).

Standard square timber is used in raises in heavy ground. Caps are used for posts in places where the ground is too heavy to permit taking out enough ground for placing a standard set.

The chute set of the raise is lined on all sides with either 2 or 3 inch lagging placed parallel with the posts.

There are two methods of bulkheading the manway when blasting: (1) A floor of 5 by 10 inch lagging is laid horizontally over the top set on the manway side, and access to the back of the raise after blasting is gained through a manhole left in the chute lining. (2) A slide of 5 by 10 inch lagging is built over the manway on the top floor, and access to the back of the raise after blasting is made by raising the top lagging.

The raise round shown in Figure 4A is used in soft ground. In medium ground another line of holes would be needed. In hard ground the "V"-cut round would be used.

Wet self-rotating and wet hand-rotated stoper machines are used in raises. One-inch quarter-octagon steel is used for Hawkesworth shanks for stoper machines.

MINING

Conditions governing the choice of mining methods never allow much latitude. The walls of the vein and the ore body itself are very heavy and will not stand open, even when supported by timber. The vein when undercut needs support as soon as an opening is made and soon develops a good deal of side and down weight.

The square-set stoping method is used throughout the mine, giving the maximum support to the walls and back, and fills the need for a flexible method of mining. Waste filling is kept as close as possible to the mining breast. Very often additional support is needed, in which case bulkheads, either solid or cribbed, are carried through from one level to another as the stoping progresses.

The manner of using the square-sets varies to suit local ground conditions. In general the ground in the Black Rock mine may be divided into three classes, as follows:

1. Ore and walls strong enough to be mined by carrying a stope upward with square-sets followed closely by waste filling.

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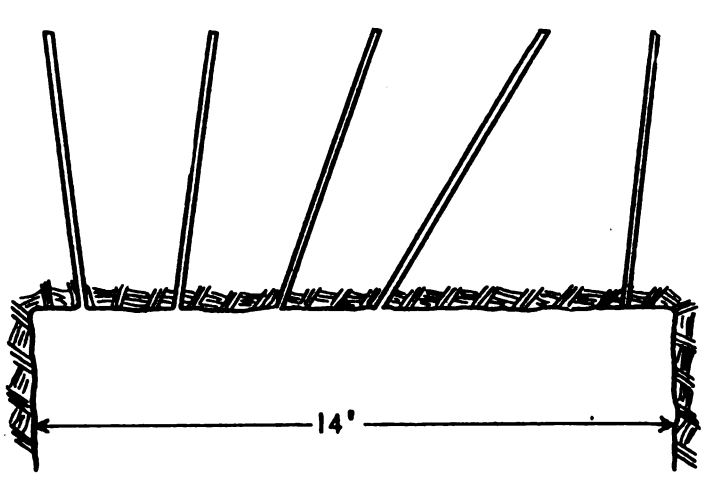

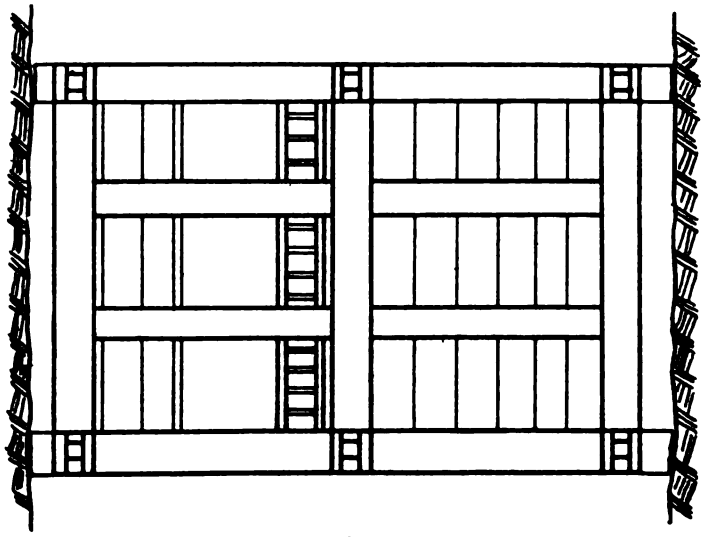

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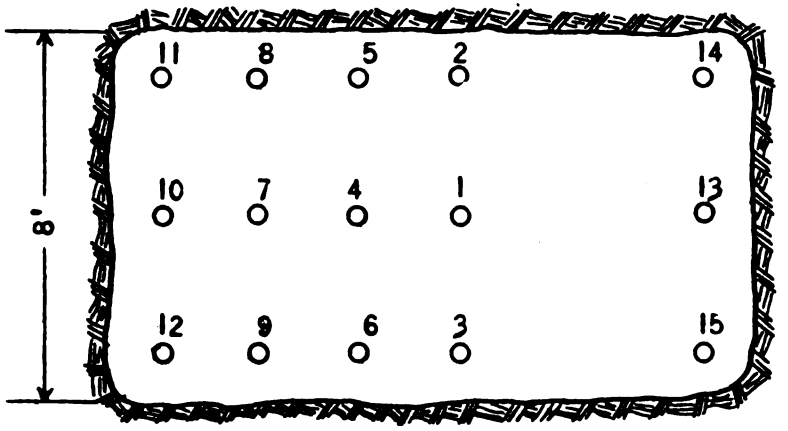

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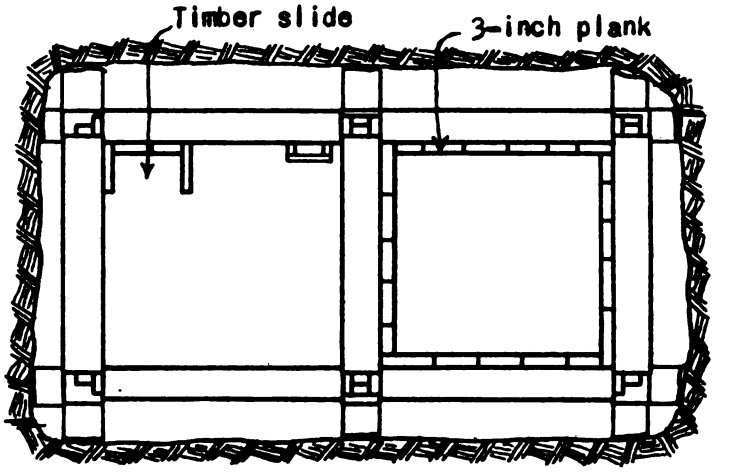

Plan

B

Figure 4.- A, Standard raise round soft ground; B, raise floor timbered

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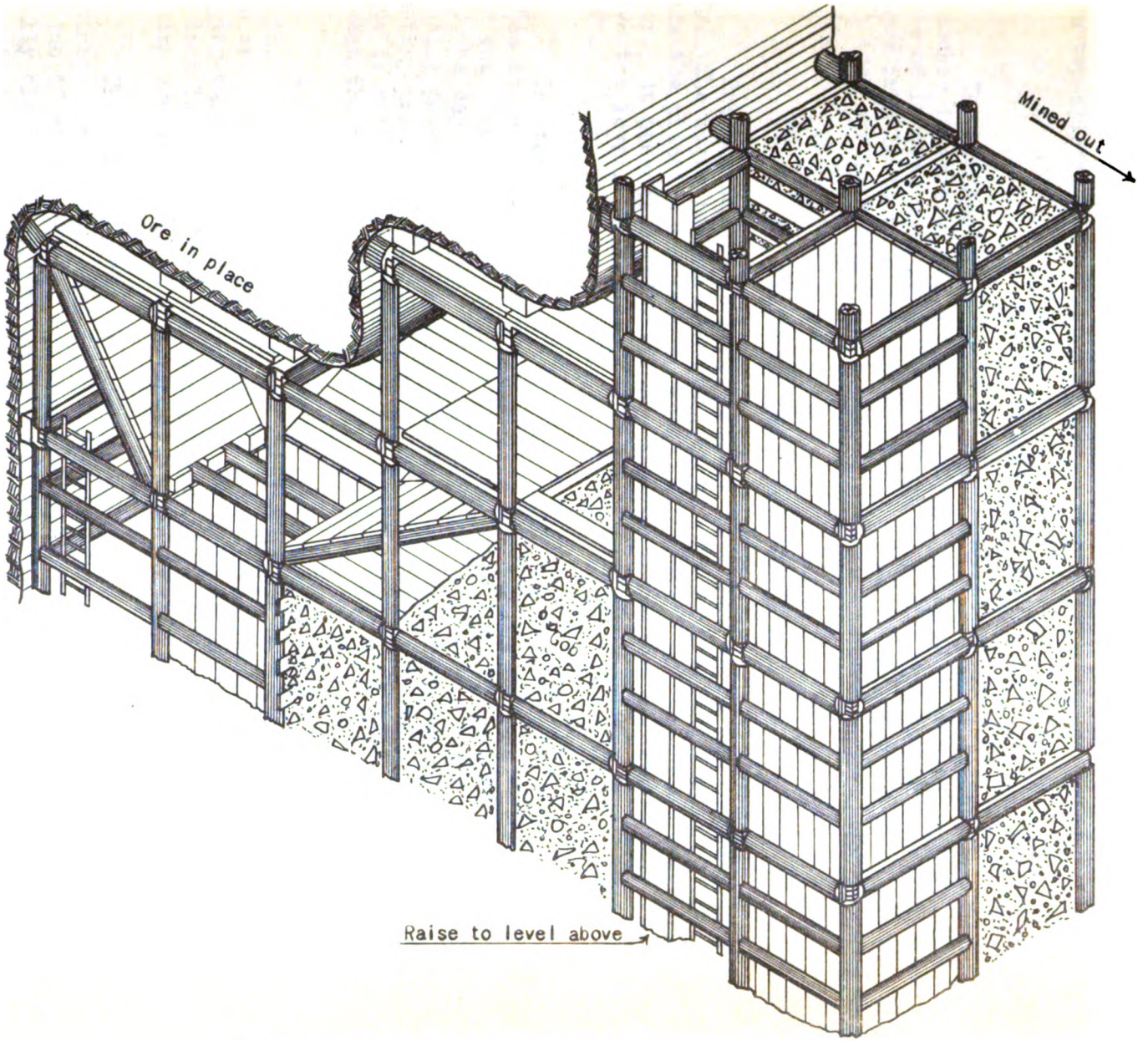

Figure 5.- Section of stope, footwall view

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2. Loose and running ground that would cave if worked upward. This class of ground is mined by underhand stoping, carrying the square-sets downward.

3. Ground intermediate between the above two classes, which is mined by a top-slicing method using square-sets.

Square-Sets

The standard square-set dimensions are 5 feet 10 inches by 5 feet 10 inches in section and 8 feet in height, measured center to center. The style of framing used is shown in Figure 6. All stope timber is round, and ranges from 10 to 12 inches in diameter. Caps and posts are framed alike, so that they are interchangeable. Occasionally, in heavy ground caps are used as posts.

Overhand Square-Setting (fig. 5)

Preparatory to stoping, a drift is driven along the footwall of the ore body. The drift is timbered with framed sets having 8-foot posts. In former years the practice has been to sill off the width of the ore body on the sill floor. At present silling off for stoping operations is done on the first floor above the level. Chutes are installed at average intervals of 30 feet along the drift.

In starting a stope "sheeting caps" are placed and blocked about 14 inches above the sill cap (fig. 3B). A short piece of lagging is placed under each end of each cap to hold it in place until the side weight is sufficient to hold the cap in position. When the second floor of the stope is completed, sheeting poles are laid across the caps and scrap timbers are placed over the sheeting poles to hold the waste fill. A similar procedure is followed if the stope is silled on the level, in which case the sheeting is carried on braces between the bottoms of the posts.

A stope is carried upward in short blocks ranging from 4 to 10 sets in length. A chute and a manway are carried at the end of the block and will in time serve as a supply raise for the next block. Where the vein is wide the hanging-wall half of the vein is stoped out first and the footwall-half later. Two-inch lagging is used to retain the waste fill on the end of the stope adjoining ore in place.

The usual practice in stoping is to blast out ground enough for one set of timbers at a time. The ore is shoveled by hand into chutes or blasted onto a slide if clean enough. The waste occurring between the bands of ore is usually blasted separately and dropped directly into the "gob." Where it is impracticable to blast the waste separately the round is blasted and held on the mining floor and sorted by hand. Extra waste needed to complete filling is obtained from the level above. Temporary flooring is laid over the entire area of each floor, and taken up before filling is run in.

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Stope chutes are lined with 2 or 3 inch plank or by "bricking," depending on the expected life of the chute. "Bricking" is the local term for a method of lining a chute by means of alternate rows of braces and short pieces of waste timber (fig. 6). A "bricked chute" will stay open longer and stand more wear from rock than a planked chute, and in heavy ground it will steady a stope as well as a bulkhead.

Stopes over two sets in width usually require bulkheads in addition to square-sets and waste filling. Usually a crib bulkhead, built with stope posts and braces, is sufficient. When the ground over a bulkhead is mined, the bulkhead is moved up a floor and waste put in its place on the floor below.

When a block has been mined to within two or three sets of the level above, a line of sets from foot to hanging wall is carried upward. The sill timbers above are caught with "stringers." The next line of sets is then mined in the same manner. The first line of sets is filled while the second line is being mined. Small poles are used to hold the waste filling in place.

When a sill floor is stoped from below, it is usually abandoned and no attempt made to fill it. If necessary, bulkheads are used to hold it open temporarily. The drifts likewise are not usually maintained after being undermined.

Underhand Mining With Square-Sets

In underhand square-setting, a raise is first put through from one level to another in the ore body to be mined. Starting from the raise a cut is taken into the ore body on the floor below the upper sill. "Booms" are then put in place, catching the bottom of the sill timbers (fig. 7). "Booms" are 5 by 10 inch timbers 13 feet long. There are two methods of holding the timbers above while the ground for the set below is being taken out: (1) If down weight is excessive, booms are used for every set. In this case two pairs of booms are used, the upper pair being moved down as mining progresses. (2) If side weight is excessive a diagonal stull will be sufficient to hold the cap in place while the ground is being worked out for posts.

After the first set is in, the ground for the set directly below is taken out. Mining progresses downward in this manner for about 50 feet. At this point an intermediate level along the footwall is established to be used for tramming ore back to the raise chute (fig. 8). Starting at the top another set is mined down. This cut is taken from the first set toward the hanging wall. Mining is continued following this procedure until a line of sets from foot to hanging wall is mined out vertically from the upper sill to the intermediate floor. The next line of sets is then mined out in the same manner. As mining progresses, waste filling is put in and held in place by small poles or broken lagging. Usually a complete column of sets is filled at one time, consisting of one set on each floor for the entire height of the stope. Two sets from top to bottom are kept open at all times, and are used for a chute and manway. One set in each line is always in use temporarily as a chute from the stope to the intermediate floor. It is not necessary to line the sets being used temporarily as chutes.

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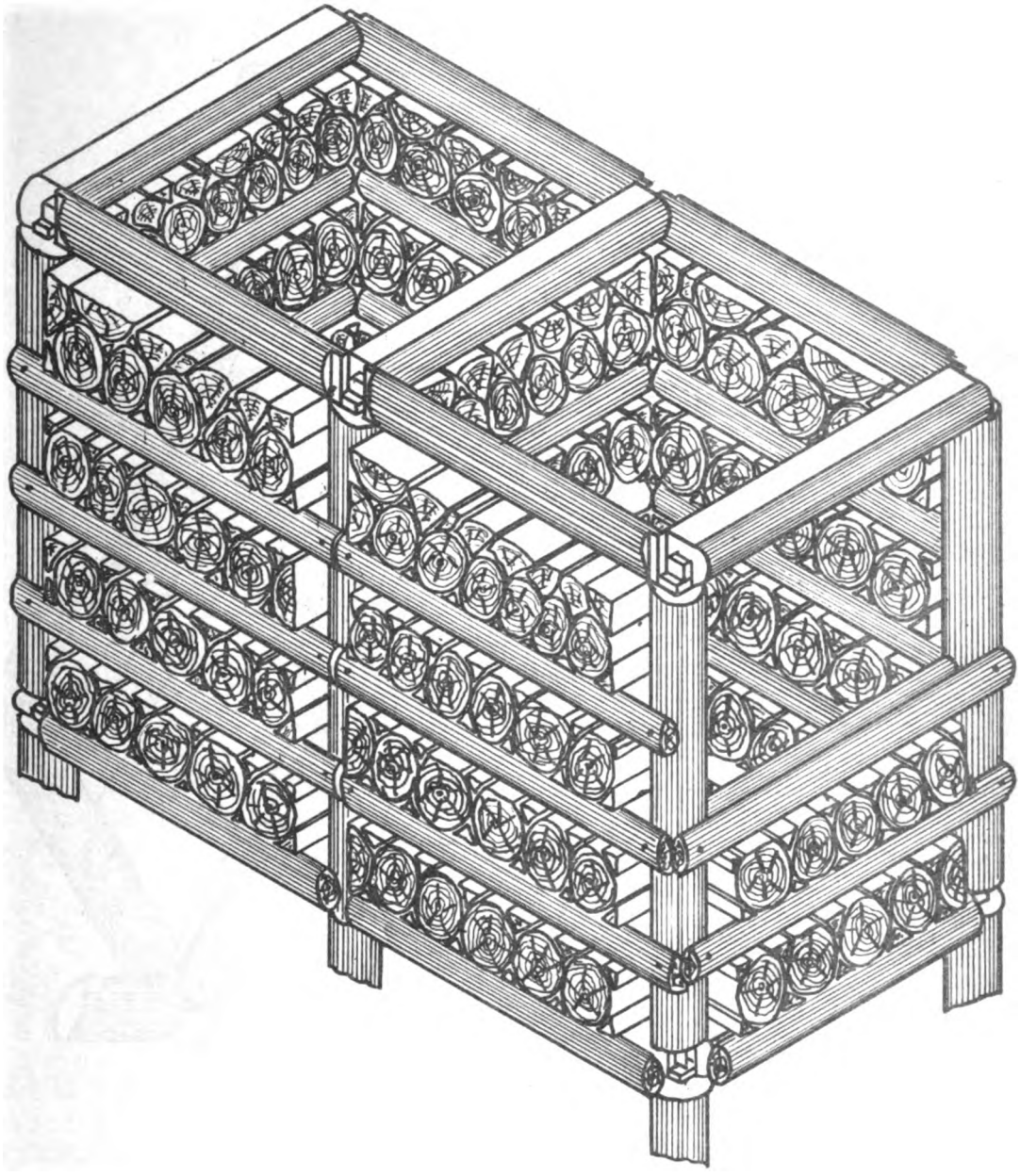

Figure 6.- Sketch showing method of lining a chute and manway with bricking

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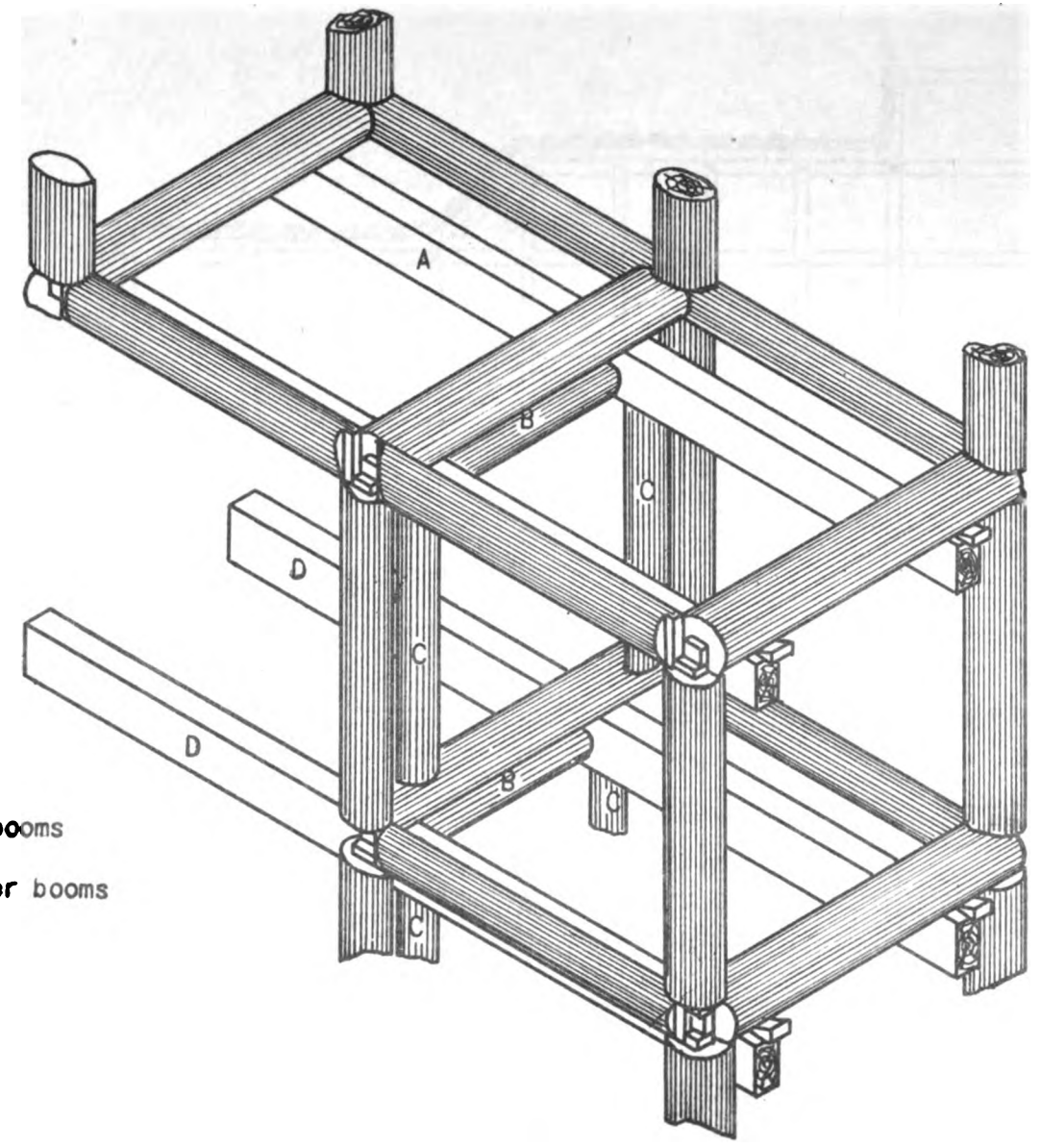

A – Upper booms

B – Braces between booms

C – Short posts under booms

D – Lower booms

Figure 7.– Sketch showing method of using booms in underhand mining

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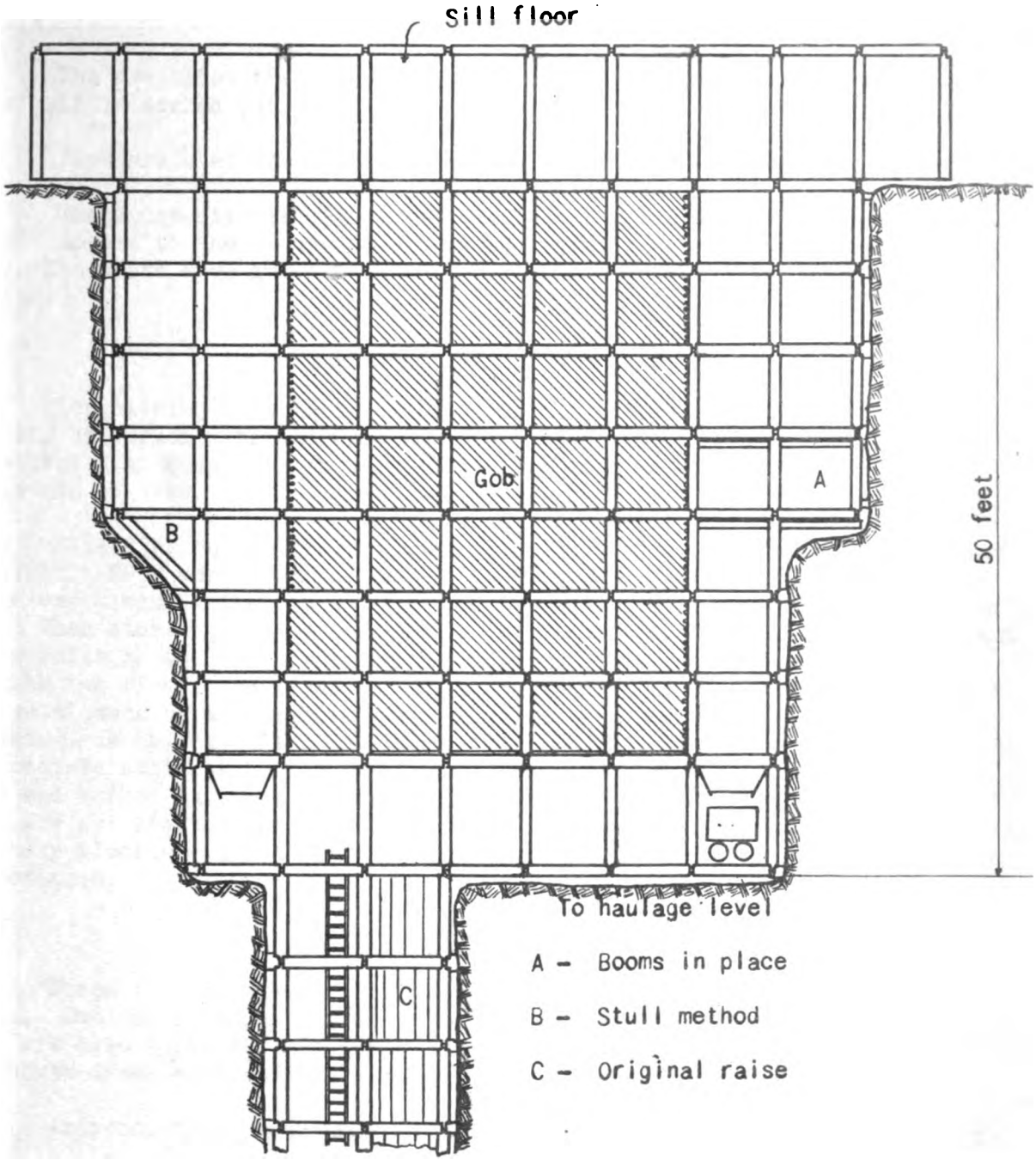

Figure 8.- Section showing underhand stopes

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As soon as a few lines of waste are put in between the original raise and the breast of the stope, a similar stope may be started in the opposite direction from the original raise. When this new stope has advanced two lines the original raise is abandoned and filled from the intermediate level to the upper sill.

The ore below the intermediate is worked in the same way after the upper half is worked out.

Caps are used for posts in underhand stoping.

The upper sill is usually abandoned before or during mining operations below. Access to the stopes is gained by footwall laterals and short crosscuts. The waste from driving these openings is conveniently used for stope filling.

Top Slicing With Square-Sets

Top slicing with square sets is used in heavy ground that would ordinarily be worked with the underhand method, except for some abnormal local conditions that makes it impractical to hold intermediate workings open for any length of time.

Depending on the ground, the slices are taken off one to three floors in height. When more than a floor is to be taken, a drift is run to the end of the ore block on the footwall of the lower floor and timbered with square sets. Then starting at the end of the ore block the floor above is mined back to the raise chute. If there are two floors of ore above the intermediate sill the two floors are mined together in retreating. Either standard sets or "cap sets" such as are employed for underhand stoping are used in this method of mining, depending on the height desired. When all the ore on and above the intermediate sill has been worked out a new slice is started below from the raise and worked in the same manner. No filling is used, and although the posts are not blasted out, the upper workings soon close in and fill themselves. Temporary flooring is used in each slice, which is recovered before the slice is abandoned.

Underground Transportation

Three-ton storage-battery locomotives are used on the main working levels. Two small storage-battery locomotives that can be moved from level to level are also employed. On levels where only a small amount of work is being done horse-drawn cars and hand-tramming are used.

Anaconda-type, end-dump, 13 cubic foot capacity cars are used on most of the levels. Two levels are equipped with side-dump, 35 cubic foot capacity cars. Maintenance of long drifts in heavy ground makes the larger cars impractical on most levels.

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The track gage is 18 inches throughout the mine. On the main haulage levels 25-pound rails are used. On intermediate levels 16-pound rails are used.

Hoisting is done in balance with skips of 7 tons capacity. Above the skips is a deck for the station tenders.

Men are hoisted and lowered on 4-deck cages, each holding eight men.

The device for changing the skips and cages is above the collar of the shaft. The cages and skips when not in use are swung onto a crawl, where they can be readily inspected and overhauled.

Rock dumped from the skip is diverted into ore or waste bins by air-operated doors.

Skip pockets are built on each level. In the upper half of the ore pocket small waste pockets are built. The capacity of the ore pockets ranges from 50 to 150 tons. Air-operated gates are used to control the flow of ore into the skips.

To catch the overflow from the skips chutes are installed in the hoisting compartments at the bottom of the shaft. From these chutes the overflow from the skips is loaded into a car and hoisted through the "chippy" compartment to the level above where it is dumped into the skip pocket.

HOISTING

The main hoist is an electrically operated first-motion double-drum hoist. Each drum is 9 feet in diameter, with a 7 foot 10 inch face. Rope speed is 2,300 feet per minute. The cable is 1½ inches in diameter, and of Lang-lay, 6 by 25 construction. The hoist installation is capable of hoisting a 7-ton skip from a depth of 5,000 feet.

The hoist is driven by a 2,000-hp., 600-volt, direct-current motor. The motor generator set consists of a 2,000-hp., 3-phase, 60-cycle, 2,200-volt motor, a steel-plate flywheel weighing 138,000 pounds, and a 1,500-kw., 600-volt, 2,500 ampere direct-current generator.

The hoist for the supply shaft is an electrically operated double-drum single-reduction hoist. The drums are 8 feet in diameter with a 7-foot face, grooved for a 1-1/8 inch rope. Hoisting speed is 1,600 feet per minute. The hoist is driven by a 600-hp., 500-volt, 960-ampere, 450-r.p.m. motor.

Both hoists are equipped with Welsh safety devices which give protection against overload, overspeed, overtravel, and provide automatic slowdown at both top and bottom of trip. The main hoist is, in addition, equipped with a Johnson trip recorder, showing time, duration, and number of trips from each level.

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FIRE HAZARDS

The large amount of timber used underground and the weight of the ground on the timber in places where electric cables are used present fire hazards that must be watched constantly. These hazards are reduced as much as possible by regular inspection of all underground working places. Conditions reported as dangerous are remedied as soon as possible.

A high-pressure water line for use in case of fire is in place in the shaft from the collar to the lowest working level, with a take-off at each station. From the take-off line there is a by-pass to the air line at each level. All stations are equipped with automatic sprinklers.

Oxygen breathing helmets are kept in good working order in the first-aid room on the surface. All shift bosses, foremen, and engineers are trained in the use of the helmets. In addition, a large number of the underground employees have had training and experience in the use of the helmets.

Exits to the adjoining Elm Orlu property are maintained on four levels.

SAFETY METHODS

The company employs a safety engineer in the mine. A trained first-aid man is on duty each shift at the collar of the shaft.

A sustained effort is made to reduce the causes of accidents in the mine. Twice a month the mine superintendent calls a meeting of safety engineer, mine foremen, and shift bosses. All conditions in the mine that might cause accidents are brought to the attention of the meeting by those present. After discussion, remedial measures are decided upon.

Shift bosses having a record of no lost-time accidents for a monthly period are given a cash bonus.

Blackboards are placed on all stations for reporting missed holes in blasting.

All chutes are covered with 10-inch round grizzlies with a maximum opening of 10 inches.

VENTILATION

The Black Rock mine is developed from three shafts, No. 1, No. 2, and No. 3 (fig. 9). No. 2 and No. 3 shafts are only 50 feet apart, and are both downcast, while No. 1 is upcast. A large centrifugal fan is used an an exhauster at the collar of No. 1 shaft. This fan has an impeller 7 feet in diameter and 3½ feet in width and runs at a speed of 350 r.p.m. It is belt-connected to a 200-hp., 575-r.p.m., 2,200-volt, alternating-current motor.

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The fan was primarily used as a pressure fan but due to the fact that the main operating shaft, No. 3, is the deepest shaft and extends to the 3,700-foot level, while No. 1 goes only to the 2,000-foot level, it is more satisfactory to have No. 3 downcast and No. 1 upcast. The ground around No. 3 shaft is very heavy, and when this shaft was upcast a tremendous amount of shaft repairs was necessary. By making No. 3 shaft downcast, considerable saving in shaft repairs has been accomplished.

When the fan was used as a blower the volume of air circulated was 150,000 cubic feet at a pressure of 4 inches water gage. Used as a suction fan the capacity is 110,000 cubic feet of air per minute. The suction as shown by the water gage is 1 inch, because the fan is getting its load much easier since there are connections on almost all of the upper levels, where mining is being done. The upper circuit in the mine extends to the 1,700-foot level; below this elevation a booster fan supplies the air.

The booster fan is 100 feet east of No. 3 shaft on the 3000-foot level. This fan has an impeller 4 feet in diameter and is a centrifugal blower with the vanes tipped backward. It is direct-connected to a 75-hp., 900-r.p.m., 2,200-volt alternating-current motor. The fan has a capacity of 65,000 cubic feet per minute.

Centrifugal fans, motor driven, having impellers 24 and 15 inches in diameter, are used for secondary ventilation. Canvas tubing 12, 16, and 24 inches in diameter is used with these fans.

Compressed-air-driven, straight-flow fans are used for ventilating drifts, raises, and dead-end stopes if the distance is not too great. Two types of compressed-air fans are used. One type has two nozzles for attachment to the compressed-air line, and the other has only one. The most valuable factor in favor of the air-driven fans is the ease with which they can be moved from place to place, and the simplicity of installation, as only a platform and a connection to the air line are required.

The size of the airways is determined by the mining conditions and not by the ventilation requirements.

Possible short-circuiting of ventilating currents is prevented by double doors forming air locks. Where the volume of traffic makes it desirable, these doors are operated by compressed-air cylinders and counterweights.

WAGE SCALE AND BONUS SYSTEM

The wages paid at the Black Rock mine are the same as are paid throughout the camp. The wage scale fluctuates, and in the last 7 years the miniumum daily wage paid underground was $4.75 and the maximum $6. The wage scale effective in the district in May, 1930, for the principal classifications of underground employees was as follows:

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Hoist engineer (surface)	$6.50	
Miner	5.25	
Mucker	5.25	
Motorman	5.25	
Pumpman	6.00	
Pipeman	5.75	
Shaftman (sinking)	5.75	(contract work)
Shaftman (repair)	5.75	
Station tender	5.75	
Timberman	5.25	
Trammer	5.25	

Practically all breaking, shoveling, and timbering operations are done on contract, as the term is used. It is really a bonus system and not contract work, as the men are given a certain definite price for the work to be done and if they make more money than the day's-pay rate they are paid the extra money as a bonus at the end of the period. If the men fall below the day's-pay rate they are guaranteed a minimum daily wage, but the particular place is investigated to see whether the failure to make bonus is due to conditions or price.

The samplers who take daily samples underground also keep a close check on all contract work and carry notebook sketches with them at all times showing a plan view of the working place. Notes are made from time to time of the progress of the work. The engineering department staff also aids in the measurement of contracts. Contracts are measured and calculated twice per month, but the bonus is paid once per month.

Prices are set by the chief contract engineer, and all contracts are checked by him at the middle and end of each month. A large sheet is kept in the office on which a plan of the work being done is sketched, and all calculations are shown so that the miners themselves can see what has been paid.

The price for drifts is generally $7 per linear foot, an over-all price that includes breaking, shoveling, and tramming. All timbering is paid for separately at $5 for each set of timber.

All timber for contract work is trucked by the miners at a price ranging from $0.50 to $1.50 per set, depending on the distance and conditions.

Nearly all raises are paid for at the rate of $0.080 to $0.100 per cubic foot of completed raise. This price is split up into $0.050 to $0.085 per cubic foot for breaking, $0.015 to $0.020 per cubic foot for timbering, and $0.015 to $0.020 per cubic foot for lining chutes.

Stope contracts are paid at a price ranging from $0.065 to $0.100 per cubic foot. The price varies, depending upon whether the rock is handled by sliding, mucking and sorting, silling, or underhand stoping. The over all price is divided into prices ranging from $0.025 to $0.040 for breaking, $0.025 to $0.040 for shoveling, and $0.015 to $0.020 for timbering.

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Other operations, such as stope filling, $3 to $6 per set; bulkheading, $5 to $10 per set; raising chutes and manways, $12 to $16 for chutes and manways; and tramming, $0.015 to $0.030 per cubic foot, are also on contract.

The wide variation in prices depends on the hardness of the ground, method of handling ore, and general conditions at the working places.

The plan of making contracts where the men buy their own powder and are paid whatever they make without the guarantee of a minimum wage has been tried.

The bonus system now being used has been successful, and it is the best plan which has been tried thus far. If the rates are equitable and fairly uniform for similar work, the miner who understands his business is paid a premium for accomplishment.

Shaft retimbering and removal of old shaft timbers are paid for at the rate of $14 to $20 per linear foot for removing old timber and putting in new.

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SUMMARY OF COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

Name of Mine: Black Rock — Period Covered: November, 1929

Tons of ore mined and hoisted: 15,929

Mining method: Square set and fill.

A. Labor (man-hours per ton):

Breaking (drilling and blasting)	0.940
Timbering and filling	1.170
Shoveling	.740
Haulage and hoisting	.870
Supervision	.208
General	.810
Total labor underground	4.738
Average tons per man shift	1.700
Labor, percentage of total cost	69.8
Average tons per man-shift on surface, properly chargeable to underground operation	5.4

B. Power and supplies:

Explosives (lbs. per ton) 30%	1.64
(lbs. per ton) 60%	.09
Timber (bd.ft. per ton, sawed stock)	12.0
(linear ft. per ton, round timber)	2.8
Total power (kw. hrs. per ton)	53.7
1. Air compression	29.8
2. Hoisting	10.3
3. Pumping	4.2
4. Ventilation	8.1
5. Lighting	1.3
Other supplies in percentage of total supplies and power	41.7
Supplies and power, percentage of total cost	30.2

C. Percentage of total cost:

Development	21.8
Stoping	78.2

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DETAIL OF DEVELOPMENT COSTS IN UNITS OF LABOR, POWER, AND SUPPLIES

	Sinking	Drifting	Cross-cutting	Raising	Total all development
Size of excavation, feet	8 by 18	7 by 9 8 by 10	6 by 8	7 by 14	
Timbered or not	Yes	Yes	No	Yes	
Physical properties of rock:					
(a) Hard or soft	Generally hard	Generally hard	Generally hard	Generally hard	
(b) Firm or loose (c) Swelling or running ground	Considerable swelling ground	Considerable swelling ground	Considerable swelling ground	Considerable swelling ground	
A. Labor (man-hours per foot)					
Breaking (drilling and blasting) Timbering Shoveling Haulage and hoisting Supervision	58.5	9.6	7.5	15.1	12.3
Total labor	58.5	9.6	7.5	15.1	12.3
Feet per 8-hour man-shift	0.14	0.8	1.1	0.53	0.65
B. Power and supplies					
Explosives (lbs. per foot)	21.0	12.5	9.9	11.8	12.3
Timber (bd.ft. per foot advance)	596	45.0	20.0	101.0	73.5
C. Labor, percent of total cost					
	- -	- -	- -	- -	68.7
Power and Supplies, per cent of total cost	- -	- -	- -	- -	31.3

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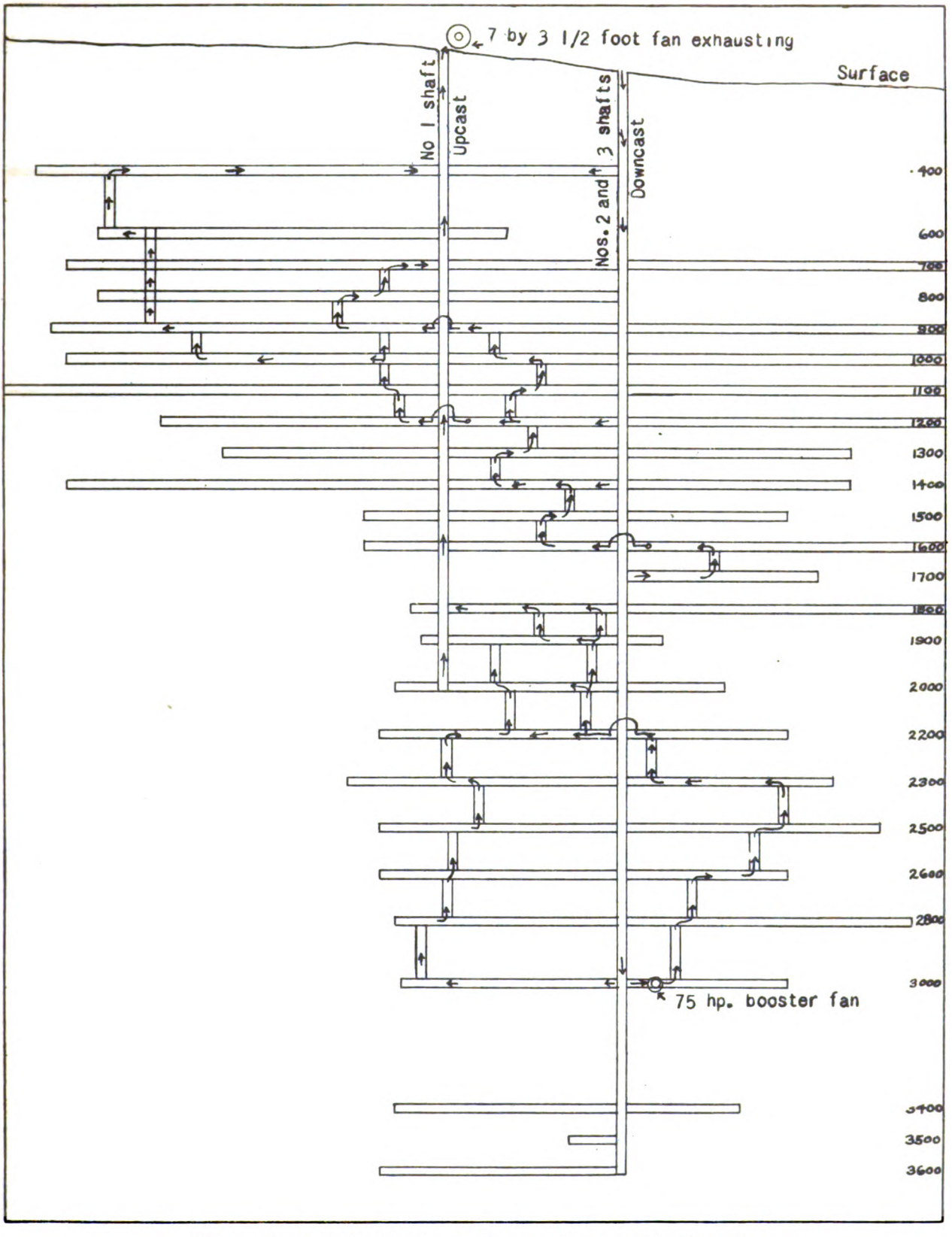

Figure 9.- Longitudinal section showing ventilation diagram

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